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MILLING METHODS AND COSTS AT THE CONCENTRATOR OF THE CANANEA CONSOLIDATED COPPER CO., CANANEA, SONORA, MEXICO.

BY

A. T. TYE

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MILLING METHODS AND COSTS AT THE CONCENTRATOR OF THE CANANEA CONSOLIDATED COPPER CO., CANANEA, SONORA, MEXICO[1]

By A. T. Tye[2]

INTRODUCTION

This paper describing the concentrator practice of the Cananea Consolidated Copper Co. at Cananea, Sonora, Mexico, is one of a series of articles on milling methods and costs being prepared by the U. S. Bureau of Mines.

The Cananea concentrator treats 2,400 tons of copper ore per day by preferential flotation, whereby the copper sulphides are floated and the iron pyrite depressed. During the first half of the year 1929 the heads to the concentrator averaged 3.29 per cent of copper, the recovery was 95.74 per cent of the total copper, and the concentrates produced contained 21.44 per cent of copper.

ACKNOWLEDGMENTS

Credit is due C. U. Coolidge, chief mining engineer, for data on mining methods; To R. V. Perry, chief geologist, for information on the character of the ore; to H. L. Lynn and Stewart Udell, of the geological department, for microscopic examination of the tailing; and to H. H. Fox, chief chemist, for method of assay of oxide copper. The assistance of Dr. T. G. Chapman, consulting engineer, U. S. Bureau of Mines, in the preparation of the paper is also acknowledged

ORE TREATED

The ore treated in the concentrator is mined from the Capote, Colorada, Veta, and Elisa mines, situated about 1 mile from the concentrator.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6261."

2 - One of the consulting engineers, U. S. Bureau of Mines and assistant concentrator superintendent, Cananea Consolidated Copper Co.

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The following tabulation gives the percentage of ore mined by various methods:

Mining method	Per cent of total ore mined
Shrinkage stopes	5.7
Square-set stopes	11.3
Rill cut and fill stopes	16.5
Horizontal cut and fill stopes	21.9
Cave stopes	22.5
Top-slicing stopes	10.2
Development	11.9
Total	100.0

The valuable metals contained in the ores are primarily copper, with small amounts of silver and gold. The ores are of about medium hardness for crushing and grinding but are entirely distinct from each other in chemical composition, and in physical and mineralogical characteristics, as is partly shown by the following tabulation:

Ore	Analysis, per cent					
	Copper	Iron	Lime	Sulphur	Silica	Alumina
Capote	1.78	15.9	0.4	14.6	44.9	10.7
Colorada	4.32	7.5	.2	6.7	58.4	11.6
Veta	2.22	7.0	.?	7.6	64.6	7.6
Elisa	2.78	16.2	16 0	7.0	31.3	6.2

Capote ore is high in iron pyrite and low in copper, and contains some hematite and magnetite. Elisa ore is high in iron but low in sulphur and has a high percentage of lime and yellow garnet. Veta ore contains large amounts of hard cherty quartz. Most of the iron and sulphur in the Colorada ore is chemically combined with copper as chalcopyrite; it also contains some iron pyrite and magnetite. The principal copper mineral in these ores is chalcopyrite, with chalcocite, bornite, and covellite occurring in minor quantities. The moisture in the crude ore averages about 4.5 per cent.

The Cananea ores have been described by R. V. Perry as follows in a letter to the author:

"The ore treated by the Cananea concentrator is variable in character because the various Cananea deposits are so different in structure and mineral content. Ore bodies in limestone contain bornite, chalcopyrite, tetrahedrite, and pyrite, in a gangue of silicified lime and garnet. The valuable mineral is localized into rich lenses and smaller blebs scattered through the silicified and garnetized limestone. Thus, while the ore is hare and resistant, the sulphides are readily liberated by crushing.

"Ore bodies in the soft altered rocks of the district are highly pyritic with values largely confined to the zone of secondary chalcocite. The sulphides are pyrite, sooty chalcocite, and a small amount of chalcopyrite. The gangue is soft altered rock containing variable amounts of quartz. Microscopic examination shows that minus 60-mesh tailings do not contain copper minerals included with the pyrite. There is a tendency for this

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high pyritic ore to oxidize as soon as broken, thus adding to the difficulties of flotation.

"Deeper level primary ores and ores from the newly-developed Colorada ore body are highly siliceous. The sulphides are chalcopyrite, bornite, tetrahedrite, tennantite primary chalcocite, molybdenite, and pyrite in a quartz and hard feldspathic rock gangue. The quartz is very brittle and is easily crushed, liberating the sulphides and permitting a high recovery of the copper minerals."

For best flotation results, ore should be mined by a method which delivers the fresh ore as soon as possible to the concentrator, and the less time it remains in the concentrator bins after crushing the greater the probability of obtaining favorable flotation results.

Ore from the upper levels of the Capote mine, which can best be described as "tarnished," is not very amenable to flotation treatment and yields poor results by flotation methods in the laboratory and in the concentrator. Chemical analyses, including determinations of oxidized copper, give very little information as to the floatability of the copper minerals. Ore from the upper levels may show very little oxide copper, but may give very low extraction by flotation, whereas freshly-broken ore from the lower levels may give excellent extraction, although chemical analysis shows some oxide copper.

BRIEF HISTORY OF CONCENTRATION

The present concentrator was built about 23 years ago. At that time gravity concentration with trommels, jigs, hydraulic classifiers, tables, and vanners constituted the method of treatment. In 1916 bulk flotation was added as an adjunct to gravity concentration.

In 1922 the concentrator was remodeled by David Cole for either straight flotation or combined flotation and gravity concentration. The main ore supply for the concentrator at that time came from the Capote mine, and as the combined iron, sulphur, and copper in the ore amounted to about 35 per cent the ratio of concentration was low, amounting to 2.57, and the copper content of the concentrate was only 4.38 per cent.

Laboratory tests run at Cananea in 1921, using coal tar and lime, proved that nearly all the iron pyrite, which at that time was being concentrated with the copper minerals and sent to the smelter for treatment, could be discarded in the tailings. These tests were run from information received from David Cole, Munroe of Granby, and Ralston of the United States Bureau of Mines. In 1923 William Wraith had the laboratory tests repeated at Cananea, Inspiration, and Tooele, using T-T reagent, cyanide, lime, and coal tar. As these tests confirmed the tests made in 1921, a pilot mill was at once installed and tests made, using standard concentrator machines. The pilot-plant tests demonstrated that the iron pyrite could be depressed. Lime was used as the depressing reagent, as suggested by Ralston in 1917. Table 1, showing the results of combined gravity and bulk flotation treatment in 1923 and the results from preferential flotation in 1929, furnishes significant figures for comparison.

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TABLE 1. - Comparison of combined gravity and bulk flotation with preferential flotation

Flow sheet	Combined gravity and bulk flotation	Preferential flotation
Period	Last half, 1923	First half, 1929
Cost of concentrating:		
Operation, per ton	$0.62	$0.51
Repairs, per ton	.16	.10
Total, per ton	.83	.69
Ore concentrated, dry tons	252,120	406,010
Ratio of concentration	2.57	6.81
Recovery:		
Copper, per cent	87.40	95.74
Silver, per cent	88.21	96.19
Gold, per cent	95.80	88.28
Power, kw.-h. per ton,	15.8	15.2
Pine oil used per ton of ore treated, pound	.61	.14
Xanthate used per ton of ore treated, pound	----	.086
Lime used per ton of ore treated, pound	----	8.80
Cost of flotation reagents, including lime, per ton	$.0954	$.0550
Copper in ore, per cent	1.95	3.29
Iron in ore, per cent	16.70	10.31
Copper in concentrates, per cent	4.38	21.44
Copper in tailings, per cent	.29	.19
Iron in tailings, per cent	5.20	6.60
Insoluble in concentrates, per cent	15.20	12.30
Moisture in concentrates, per cent	11.40	9.80
Cost of rods and balls per ton of ore treated	$.0790	$.0690
Copper produced in concentrates, tons	4,286	12,777
Plus 48-mesh material in tailings, per cent	4.1	3.0
Plus 65-mesh material in tailings, cumulative per cent	15.6	14.1
Minus 200-mesh material in tailings, per cent	61.1	60.2

Coarse breaking and intermediate crushing

There are eight crude-ore bins at the crushing plant (see fig. 1), which hold about 2,500 tons of ore. Theore is taken from the bins by a traveling feeder, which delivers it to a 30-inch belt running at the rate of 157 feet per minute, discharging into the hopper of a No. 8 type N Allis-Chalmers gyratory crusher. Grizzlies are not used before the crusher to screen out the fines. The maximum size received by the gyratory crusher is governed by the mine grizzlies, with openings spaced about 15 inches apart. The gyratory crusher is driven by a 75-hp. motor, the driving shaft operating at 450 revolutions per

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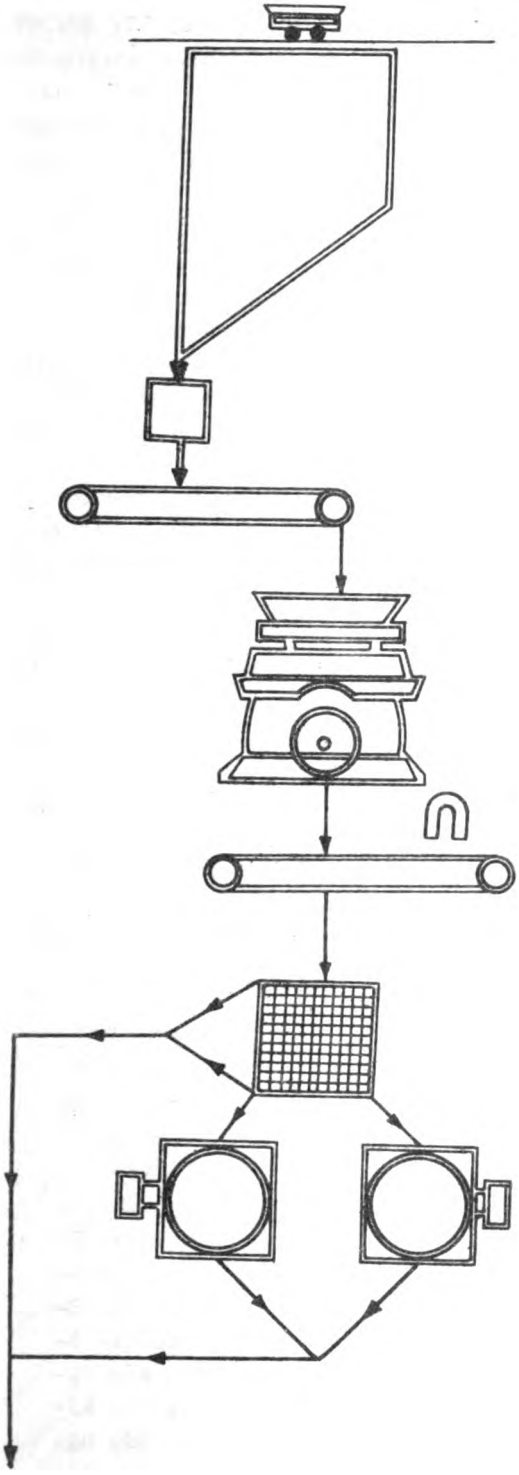

Railroad cars from mines

Eight crude-ore storage bins
2,500 tons capacity

Conveyor belt, 30-inch, 157
feet per minute. Run-of-mine
ore, 15 inches

No. 8, type N, Allis Chalmers
gyratory crusher, 75-hp. motor

Dings suspended magnet

Dings head-pulley magnet

Conveyor, 30-inch, feed
3 1/2 inches maximum size,
220 feet per minute

Cole shaking screen, 44 by
39 inch, 430 shakes per
minute, 1-inch opening

Two 48-inch Symons vertical
disk crushers

Figure 1. - Flow sheet of crushing plant

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minute. The ore is crushed to about 3 1/2-inch maximum size and delivered to a 30-inch conveyor, traveling 220 feet per minute, which passes under a Dings suspended magnet and over a Dings head-pulley magnet, which removes all the tramp iron. These magnets have proved invaluable, and iron rarely passes them. The conveyor delivers the product to a 44 by 49 inch Cole shaking screen operating at 430 revolutions per minute. The screen is made of 1/4-inch double crimped wire and has 1-inch openings. The entire crushing plant tonnage passes over this screen, and although the screen was designed to treat 1,200 tons per day it treats at present 2,600 tons per day, and has proved to be simple and satisfactory. The oversize of the Cole screen is crushed by two 48-inch Symons fine-reduction vertical disk crushers with driving shafts operating at 300 revolutions per minute, driven by 100-horsepower motors. The discharge of the Symons crushers with the undersize of the Cole screen is the concentrator feed. The crushing-plant product is conveyed to the fine-ore bins, of 2,400 tons capacity, by a 24-inch conveyor running 570 feet per minute on a slope of 4 1/2 inches per foot.

The present crushing plant, as remodeled by David Cole in 1922, was designed to crush 200 tons of ore per hour, so that the entire daily concentrator tonnage of that time could be produced on one shift. Since that time the concentrator capacity has been increased to 2,400 tons per day, so that the crushing plant, although operating at a capacity of 220 tons of ore per hour, must run overtime. The crushing plant will probably be remodeled at an early date and Symons cone crushers installed. Dust is removed from the air of the crushing plant by an exhaust fan, which deposits the dust in an iron tank, from which it is returned to the ore going to the fine-ore bins.

The products of the various crushing machines vary widely, depending upon the character of the ore. Table 2 gives screen sizes of the various intermediate and final products of the plant when operating on Colorada ore.

TABLE 2. - Screen sizes of crushing plant heads and products, per cent

Size	Mine run	Gyratory discharge	Cole screen		Symons discharge
			Oversize	Undersize	
+3 inch	53.7	14.2	29.4	-	-
-3 +2 inch	4.4	15.6	25.3	-	-
-2 +1 inch	8.4	22.9	20.2	-	25.9
-1 + 1/2 inch	10.5	10.8	11.9	21.6	49.4
-1/2 inch +3 mesh	4.6	9.9	3.4	16.2	10.9
-3 +4 mesh	2.4	3.1	1.4	9.1	2.6
-4 +6 mesh	1.8	2.6	.9	6.5	1.8
-6 +8 mesh	1.9	2.6	.9	6.5	1.5
-8 +10 mesh	1.4	2.1	.7	5.6	1.1
-10 +14 mesh	1.2	1.9	.6	4.5	.9
-14 +20 mesh	1.0	1.4	.5	3.6	.7
-20 +28 mesh	1.0	1.4	.5	3.2	.6
-28 +35 mesh	.8	1.4	.5	2.8	.6
-35 +48 mesh	.7	1.0	.4	2.3	.5
-48 +65 mesh	.7	1.0	.4	2.1	.4
-65 +100 mesh	.6	1.0	.4	1.9	.4
-100 +150 mesh	.4	.7	.3	1.3	.3
-150 +200 mesh	.4	.5	.3	1.0	.3
-200 mesh	4.1	5.9	2.0	11.8	2.1
Total	100.0	100.0	100.0	100.0	100.0

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Table 3 shows the distribution of oxide copper in Capote and Veta ores and the distribution of oxide copper in the material from the dust collector of the crushing plant.

TABLE 3. - Distribution of oxide copper in Capote and Veta ores and dust

Size	Oxide copper, per cent		
	Capote ore	Veta ore	Dust
1.05 inches to 14 mesh	0.02	0.03	
Through 14 mesh, on 20 mesh	.02	.07	
Through 20 mesh, on 28 mesh	.07	.07	
Through 28 mesh, on 35 mesh	.10	.07	
Through 35 mesh, on 48 mesh	.17	.07	
Through 48 mesh, on 65 mesh	.20	.08	
Through 65 mesh, on 100 mesh	.18	.12	0.06
Through 100 mesh, on 150 mesh	.20	.17	.10
Through 150 mesh, on 200 mesh	.21	.20	.10
Through 200 mesh, on 250 mesh	.24	.20	.12
Through 250 mesh, on 325 mesh	.25	.20	.16
Through 325 mesh	.30	.31	.22

Concentrator sections

The concentrator contains three sections. Sections 1 and 2 are the original drag-belt classifier sections remodeled in 1922. Section 3, the newest section, shown in Figure 2, was placed in operation the first part of 1929. In sections 1 and 2 the ore is ground in a primary rod mill and the rod-mill discharge is sent to two Hardinge ball mills in closed circuit with Cole triplex drag-belt classifiers. In section 3 the primary rod mill is in closed circuit with a simplex classifier, which returns the coarse sand to the rod mill for additional grinding. The overflow of the simplex classifier discharges to a bowl classifier, which removes as overflow nearly all the material already ground to flotation size, and the sand is divided between two ball mills in closed circuit with Dorr classifiers.

Primary grinding in section 3

The product of the crushing plant is stored in bins of 750 tons capacity per section. A pan feeder delivers the ore to an 18-inch conveyor belt traveling 83 feet per minute, the ore being weighed automatically by a Merrick weightometer. The conveyor delivers the product to one 6 by 12 foot Cole rod mill operating at 16 revolutions per minute and loaded with 22 tons of 3 1/4-inch by 12-foot rods each weighing 333 pounds. The rods are made at Monterrey, N. L., Mexico, and have the following composition:

C	Mn	Cr	P	S	Si
0.87	0.62	0.25	0.020	0.050	0.12

The rod consumption is 0.603 pound per ton of ore milled.

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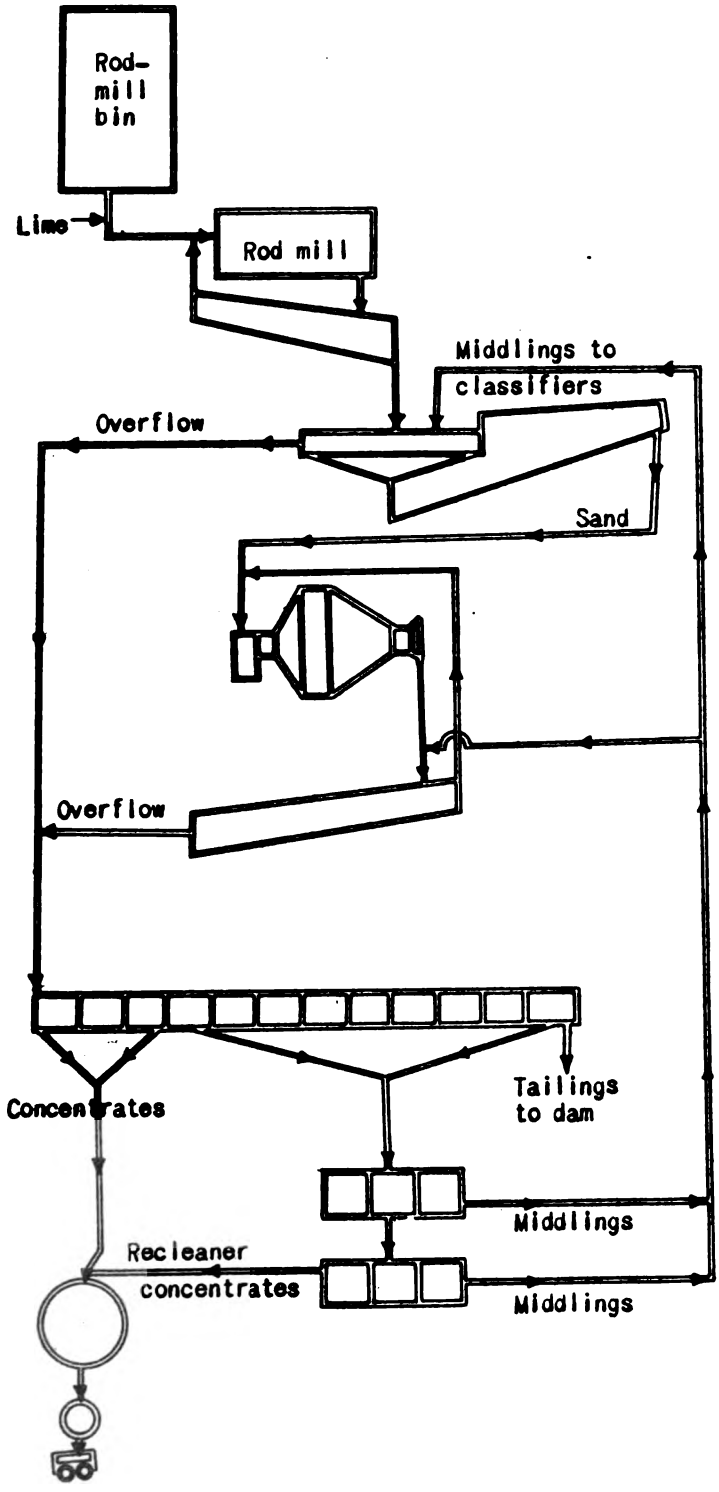

Rod-mill bin, 750-ton

Automatic lime feeders

Cole Bergman rod mill, 6 by 12 feet, 16 r.p.m., 83 kw., 22 tons of rods

Dorr classifier, 4 by 21 2/3 feet, in closed circuit with primary rod mill, Texrope drive

Bowl classifier, 6 by 28 1/3 by 12 feet, slope 2 inches, 6 r.p.m. reciprocating rakes, 24 drags per minute, Texrope drive

Two Hardinge ball mills, 8-foot, by 28-inch, 18 r.p.m., 15 tons, 2 1/2-inch balls, Texrope drive

Two DSD circulating classifiers, 6 by 23 1/3 feet, slope 3 1/4 inches, 24 drags per minute, Texrope drive

Inspiration-type flotation machine, 12 cells each. Each cell 48 by 32 inches, 1-3 cells-finished concentrates, 4-12, to cleaners

Three cleaners with 3 cells each

Recleaner with 3 cells

Two Dorr thickeners, 80-foot

Four Oliver filters, 12 by 12 feet

Railroad cars direct to roof of reverberatory furnaces

Figure 2.- Flow sheet of section 3 of the concentrator

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The rod mills are first lined with wood 1 1/2 inches thick. Then a cast-iron lining made at the company's local foundry is added. A set of cast-iron breast and end liners weighs 29,647 pounds and lasts about 10 months. The scrap is returned to the foundry. The actual liner consumption is 0.045 pound per ton of ore milled, which amounts to 0.133 pound of liner installed per ton of ore. The concentrator is charged 4 cents per pound for the lining material.

The mills are driven by 150-hp. motors through Lenox drives and harringbone gears and pinions and are supported near the discharge end by a roller bearing with six S. K. F. Universal roller bearings in the chain. The roller bearing support, together with a high pulp discharge, gives the remarkably low power input of 83.3 kw.

The tonnage ground in the rod mill is limited by the capacity of the secondary grinding section. The rod-mill capacity is limited to 700 tons per day when two ball mills are operated as secondary mills per primary rod mill. With four ball mills running as secondary grinders per primary rod mill the primary mill has a capacity of 1,200 tons or more per day. Operating time of the rod mills amounts to 98 per cent.

Secondary grinding

The Dorr classifier in closed circuit with the primary rod mill discharges into a bowl classifier, and the sands from the bowl classifier are divided evenly between two 8-foot by 28-inch Hardinge ball mills operated by Texrope drive at 18 revolutions per minute. The Hardinge mills are each in closed circuit with a Dorr classifier. The ball load of each mill amounts to 15 tons of 2-1/2-inch balls made at Monterrey and of the following composition:

C	Mn	Cr	P	S	Si
1.06	0.687	0.338	0.020	0.034	0.076

The ball consumption amounts to 1.36 pounds per ton of ore. Liners are made of manganese steel about 1 inch thick and last about 13 months. Liner consumption is 0.1346 pound per ton of ore. Lifting bars are used.

Effect of ball load on grinding

In sections 1 and 2 the Hardinge mills are operated at 16-1/2 revolutions per minute, and each mill has a ball load amounting to 9 tons whereas in section 3 the Hardinge mills operate at 18 revolutions per minute and have a ball load of 15 tons. Table 4 summarizes the results obtained under each condition and shows that the secondary mills using a 15-ton ball load actually ground 67 per cent more material through 48-mesh size, as compared to the mills operating with a 9-ton ball load, and consumed only 27 per cent more power.

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TABLE 4. - Comparison of secondary ball-mill performance in sections 1 and 3

Section number	1 and 2	3
Type of classifier in closed circuit with ball mill	Drag belt	Bowl and circulating
Speed of mills, revolutions per minute	16-1/2	18
Solids in ball mill discharge, per cent	68	68
Ball load per mill, tons	9	15
Ore per section, tons	718	1015
Ore ground through 48-mesh size, tons	134.6	224.8
Per cent of feed ground through 48-mesh size	37.5	44.3
Ore ground through 48-mesh size, per hour, tons	11.21	18.73
Power to each mill, kw.	68.0	86.4
Kw.-h. per ton of original crude ore	4.54	4.08
Kw.-h. per ton actually ground through 48-mesh size	12.12	9.23
Per cent increase in tons of ore ground through 48 mesh		67
Per cent increase in power used		27

Classification

The drag classifier operating in closed circuit with the primary rod mill of section 3 is 4 feet by 21 feet 8 inches in size and has a slope of 3 3/4 inches per foot. It is operated at 35 strokes per minute by a Texrope drive driven by a 5-hp. motor.

The drag classifiers operating in closed circuit with the secondary ball mills are 6 feet by 23 feet 4 inches and have a slope of 3 1/4 inches per foot. They are run at 24 strokes per minute by Texrope drives operated by 5-hp. motors.

The bowl classifier is 6 feet by 28 feet 4 inches, with a bowl 12 feet in diameter. The reciprocating rakes are operated by a 5-hp. motor at 24 strokes per minute and the bowl rakes by a 3-hp. motor at 6 revolutions per minute. The classifier has a slope of 2 inches to the foot. A large quantity of middlings pulp is used as back-wash water in the bowl classifier, and a spray of water keeps the fines washed out of the sands.

Nearly one-half of the feed to the section is overflowed by the bowl classifier as finished product to the flotation machines. Besides the regular mill feed, which is overflowed by the bowl and circulating classifiers, all the flotation middlings pulp is returned to these classifiers as make-up water. As the middlings pulp contains about 300 tons of solids per 1,000 tons of original feed these three classifiers overflow 1,300 tons per 1,000 tons fed to the primary rod mill.

Table 5 gives screen sizes of feed to primary rod mill and screen sizes of the sand and overflow products of the classifier in closed circuit with the primary rod mill.

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TABLE 5. - Screen sizes of heads and products of primary grinding circuit

Size	Heads to rod mill	Classifier in closed circuit with rod mill	
		Sand	Overflow
On 1 inch	4.7		
On 1/2 inch	33.2		
On 3 mesh	15.5		
On 4 mesh	5.9		
On 6 mesh	4.6	1.0	
On 8 mesh	4.4	3.2	
On 10 mesh	3.9	8.7	0.6
On 14 mesh	3.5	14.6	2.9
On 20 mesh	2.6	21.0	6.8
On 28 mesh	2.7	21.1	9.8
On 35 mesh	2.4	13.4	10.0
On 48 mesh	1.9	6.5	8.7
On 65 mesh	2.0	3.8	8.7
On 100 mesh	1.8	2.1	7.6
On 150 mesh	1.4	1.0	5.2
On 200 mesh	.8	.5	3.2
Through 200 mesh	8.7	3.1	36.5
Total, per cent	100.0	100.0	100.0

Table 6 gives screen sizes of heads to secondary grinding circuit and screen sizes of finished and intermediate products of this circuit. The middlings of the flotation treatment are normally returned to the classifiers of the secondary grinding circuit and as these middlings form a large portion of the classifier overflows and since approximately 85 per cent of the middlings is minus 200-mesh size the screen analyses under these conditions are misleading. The screen sizes of Table 6 are given for July, 1929, and show the actual size produced by grinding, as middlings were not returned to the classifiers for the period given.

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TABLE 6. - Screen sizes of heads, intermediate, and final products of secondary grinding circuit

Size	Bowl-classifier heads	Bowl-classifier sand	Ball-mill discharge	Circulating classifier		Bowl-classifier overflow	Combined overflows
				Sand	Overflow		
On 8 mesh	0.20						
On 10 mesh	1.22	2.22					
On 14 mesh	4.24	7.46	0.50	0.76			
On 20 mesh	7.92	14.06	1.58	1.84			
On 28 mesh	10.40	18.18	4.32	5.10			
On 35 mesh	9.28	17.58	8.06	9.80			
On 48 mesh	9.06	13.92	14.58	15.44	1.68	1.32	0.84
On 65 mesh	8.44	11.66	21.58	23.98	8.62	3.40	4.20
On 100 mesh	6.92	7.28	18.04	18.12	14.00	6.20	8.78
On 150 mesh	4.98	3.36	12.92	12.58	10.50	7.02	8.64
On 200 mesh	3.08	1.24	6.12	5.52	11.82	5.54	6.60
Through 200	34.26	3.08	12.30	6.86	53.38	76.52	70.94
Total, per cent	100.00	100.00	100.00	100.00	100.00	100.00	100.00

Table 7 gives chemical analyses of classifier products produced in the grinding circuit.

TABLE 7. - Chemical analyses of classifier products

Product	Copper, per cent		Iron, per cent	Sulphur, per cent	Lime, per cent	Insoluble, per cent
	Total	Oxide				
Rod-mill classifier sand	6.14	0.03	17.6	6.14	0.2	54.0
Rod-mill classifier overflow	3.68	.03	10.6	9.20	.2	72.4
Ball-mill classifier sand	5.76	.03	19.4	5.76	.2	52.2
Ball-mill classifier overflow	4.36	.03	10.4	11.50	.3	69.8
Bowl-classifier sand	4.44	.03	10.4	10.80	.3	71.3
Bowl-classifier overflow	3.24	.02	6.8	6.5	.3	76.4

Table 8 gives a screen test with chemical analyses of the various screen sizes of the ball-mill classifier sand, and Table 9 gives a screen test with analyses of the screen sizes of the bowl-classifier sand.

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TABLE 8. - Sizing and chemical analyses of ball-mill classifier sand

Mesh	Weight, per cent	Copper, per cent		Iron, per cent	Sulphur, per cent	Lime, per cent	Insoluble, per cent	Combined iron copper sulphur, per cent
		Total	Oxide					
On 14	0.90	1.66	0.04	6.00	2.9	0.2	82.6	10.56
On 20	2.38	1.92	.05	5.00	3.3	.1	82.6	10.22
On 28	5.68	2.54	.03	6.40	5.1	.1	79.2	14.04
On 35	10.60	3.52	.04	8.00	8.7	.1	73.6	20.22
On 48	15.82	4.44	.03	12.60	13.0	.2	64.4	30.04
On 65	21.48	6.20	.05	19.80	21.2	.2	50.0	47.20
On 100	17.20	7.76	.04	30.40	33.1	.1	26.8	71.26
On 150	11.56	7.02	.04	33.80	37.7	None.	17.6	79.42
On 200	7.24	8.00	.04	34.40	38.6	None.	17.2	81.00
Through 200	7.14	7.92	.09	22.00	24.8	.5	41.0	54.72

TABLE 9. - Sizing and chemical analyses of bowl-classifier sand

Mesh	Weight, per cent	Copper, per cent	Iron, per cent	Sulphur, per cent	Lime, per cent	Insoluble, per cent	Combined iron copper sulphur, per cent
On 10	3.88	2.02	6.4	5.3	0.3	79.2	13.7
On 14	10.32	1.40	4.2	3.2	.3	85.2	8.8
On 20	16.46	2.14	5.6	5.3	.3	81.8	13.0
On 28	18.94	3.28	10.0	10.3	.3	69.8	13.6
On 35	16.94	4.42	13.8	15.1	.2	61.4	33.3
On 48	12.40	4.90	14.8	15.9	.3	58.6	35.6
On 65	9.98	5.92	13.6	17.1	.2	56.0	36.6
On 100	5.98	6.62	16.4	18.2	.3	54.6	41.2
On 150	2.10	8.72	19.4	22.7	.2	44.8	50.8
On 200	1.10	8.62	22.0	25.6	.3	38.0	56.2
Through 200	1.90	4.24	12.8	13.8	2.3	56.8	30.3

The concentration of the sulphide minerals in the classifier sand products may be noted in Table 7. The bowl classifier overflows, the lighter material leaving the heavy sulphides in the bowl sands for additional grinding. The bowl-classifier sands contain 10.4 per cent iron, whereas the ball-mill classifier sand contains 19.4 per cent iron, and these analyses explain the larger proportion of fines in the ball-mill olassifier sand as compared to the bowl-classifier sand, as shown in Tables 8 and 9. The combined iron, copper, and sulphur content in the finer sizes of the ball-mill classifier sand may amount to 80 per cent of the total weight of these sizes, leaving insoluble approximately 17 per cent.

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Table 10 gives screen analyses of ball-mill classifier overflow products, showing the variation in the products with varying percentages of solids. Table 11 gives similar data for the bowl-classifier overflow products.

TABLE 10. - Screen sizes of ball-mill classifier overflow products in per cent, with varying percentages of solids

	Solids, per cent									
Mesh	20.3	21.9	23.3	23.5	24.3	25.2	25.3	26.9	27.2	32.7
On 48	0.64	0.86	1.84	1.54	1.30	1.30	2.66	2.20	3.40	4.12
On 65	5.30	6.04	7.30	8.62	7.80	7.82	6.44	8.68	12.40	11.74
On 100	11.70	12.16	11.88	13.34	12.84	13.30	10.56	13.58	14.30	13.82
On 150	12.10	9.44	10.56	12.02	10.82	12.42	9.40	12.80	10.54	10.30
On 200	8.30	11.76	11.58	7.92	8.02	8.24	11.26	7.88	11.28	11.66
Through 200	61.96	59.74	56.84	56.56	59.22	56.92	59.68	54.86	48.08	48.36

TABLE 11. - Screen sizes of bowl classifier overflow products in per cent, with varying percentages of solids

	Solids, per cent						
Mesh	19.8	22.4	22.9	23.1	24.8	25.0	25.7
On 48	0.54	1.32	1.22	2.20	3.16	4.18	4.26
On 65	2.48	3.40	4.10	4.22	5.10	5.46	6.38
On 100	5.42	6.20	6.40	6.64	6.92	7.64	8.90
On 150	6.72	7.02	6.68	6.90	6.64	6.84	6.34
On 200	7.40	5.54	7.64	5.32	7.20	6.88	6.64
Through 200	77.44	76.52	73.96	74.72	70.98	69.00	67.48

Flotation

The Inspiration type of flotation machine is used at Cananea. The Forrester, McIntosh, and Cole-Bergman machines have been used, but the Inspiration type appears to be a little better than the others for the peculiar water and ore conditions at Cananea. The flotation department has 8 Inspiration roughing flotation machines, each of 12 cells, measuring 48 by 25 inches. The free canvas area in each machine amounts to 106 square feet. The machines have a slope of one-half inch to the foot, which is necessary due to the coarse sand and heavy iron pyrite in the pulp. A cleaner machine of 3 cells follows each rougher machine of 12 cells. The concentrate from 2 cleaner machines is again cleaned in a machine of 3 cells.

National No. 3 woven blankets have been found best suited to local conditions. The blankets choke very rapidly, because the return water from the tailings dam is saturated with calcium sulphate and a gelatinous material precipitates from the soluble salts of the ore when the ore is brought in contact with the lime in the mill pulp. Several men are required each shift for punching blankets, otherwise the blankets become quite dead. Pieces of rubber 5 1/2 by 2 1/2 by 1 1/2 inches attached to wooden rods are used for punching. The life of blankets is approximately 22 days. Rubber blankets have been tried and have

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been found unsatisfactory because lime salts deposit a horny cake and cover the holes of the blanket.

One No. 8 Roots blower, 30 by 52 1/2 inches in size, running at 170 revolutions per minute, furnishes 10,100 cubic feet of air per minute at a pressure of 5 1/4 pounds, which is sufficient for seven or eight flotation roughing machines and cleaners. The blower is driven by an Allis-Chalmers synchronous motor using a Texrope drive. The motor requires approximately 209 kw. and furnishes air for 2,500 tons of ore per day, the power consumption for flotation air amounting to 2 kw.h. per ton of ore treated.

Dry lime is added to the rod-mill feed at the rate of 5.0 pounds per ton of ore by automatic belt feeders, which give a very uniform alkalinity amounting to about 0.56 pound of CaO per ton of water in the flotation pulp. A milk of lime plant is being installed. Potassium xanthate is used at rate of 0.086 pound per ton of ore and is added just before the heads of the rougher cells. If added in the grinding circuit it causes coarse-ore particles to overflow the classifier, increasing the tailings assay and reducing the grade of concentrate in the flotation treatment. Yarmor pine oil is added to classifier overflows at the rate of 0.14 pound per ton of ore treated.

Each flotation roughing machine of 12 cells treats approximately 350 tons of ore per day. The concentrates from the first three cells of the rougher are finished product and go directly to thickeners. This product amounts to 70 to 80 per cent of the total concentrates. The remaining nine cells of the rougher produce concentrates that are cleaned and recleaned in cleaner machines, the concentrates from the final cleaner machine joining the concentrates of the first three cells of the rougher and going to thickeners.

Table 12 gives chemical analyses of flotation head and intermediate and final products of the flotation treatment. The concentrator feed, as indicated by the rod-mill classifier overflow, was 2.89 per cent copper, which mixed with the combined flotation middling returned to the classifiers of the grinding circuit and containing 0.93 per cent copper gave flotation heads containing 2.44 per cent copper. The last nine cells of the rougher machine produced concentrates containing 0.93 per cent copper, this product being cleaned to products analyzing 5.36 and 12.66 per cent copper in the first and second cleaner machines respectively.

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TABLE 12. - Chemical analyses of flotation heads and products

	Analyses, per cent				
	Copper	Iron	Sulphur	Lime	Insoluble
Rod-mill classifier overflow	2.89	8.8	9.2	0.4	71.2
Combined middlings to classifiers	.93	8.8	9.1	.6	73.2
Flotation heads	2.44	8.0	8.6	.5	73.8
Concentrates from rougher cells 1, 2 and 3	23.42	26.0	31.2	.2	14.4
Concentrates from rougher cells 4 to 12, inclusive	.93	8.4	8.6	.5	72.0
Flotation tailings from rougher	.15	6.4	6.3	.5	80.8
Concentrates from first cleaner machine	5.36	18.2	20.1	.3	49.0
Middlings from first cleaner machine	.68	8.0	8.6	.5	73.8
Concentrates from second cleaner machine	12.66	31.8	37.0	.2	14.6
Middlings from second cleaner machine	2.37	13.6	15.1	.3	60.8
Combined flotation concentrates	21.10	25.2	30.3	.2	17.4

Table 13 gives screen sizes of the flotation heads and the intermediate and final products of flotation treatment.

TABLE 13 - Screen sizes of flotation treatment products, per cent

Mesh	Rod-mill classifier overflow	Combined flotation middlings	Flotation heads	Concentrates from rougher cells 1, 2 and 3	Concentrates from rougher cells 4 to 12 inclusive	Tailings from rougher cells	Concentrates from first cleaner machine	Middlings from first cleaner machine	Concentrates from second cleaner machine	Middlings from second cleaner machine	Combined concentrates
On 10	4 66										
On 14	7 30										
On 20	9.62										
On 28	10.20										
On 35	9.06										
On 48	7.00	0.14	0.38	0.54	0.08	0.50	0.10	0.12	0.68	0 18	0.18
On 65	7.34	.82	3 58	2.04	.72	7.66	1.18	.58	2.76	1.10	1 46
On 100	5.96	2.38	8.86	4.96	1.84	13 22	4 38	1.50	9.04	3.08	6.16
On 150	3 92	2.90	8.68	11.18	2.76	9.66	6.40	2.76	12.86	5.06	10.48
On 200	2.78	2.18	5.34	15.10	3.06	5.92	5.52	3.86	11.26	5.62	10.60
Through 200	32.16	91.58	73.16	66.18	91.54	63.04	82.42	91.18	63.40	84.96	71.12

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DEWATERING AND HANDLING OF CONCENTRATES

The combined flotation concentrates from the three concentrator sections are elevated by one of two Wilfley pumps and delivered to one of two 80-foot Dorr thickeners. The concentrates pulp, as it enters the thickener, contains 15 to 18 per cent of solids, is at a temperature of 69°F., and is alkaline to the extent of 0.127 pound of CaO per ton of water.

The overflow water of the thickener is returned to the head of the mill by one Cameron 2-stage centrifugal pump driven by a 25-hp. motor. The thickener spigot discharge contains 70 per cent of solids and goes to two of four 12 by 12 foot Oliver drum filters operating at one revolution in 14 minutes.

The pulp in the filter, which is maintained at a level of 51 inches below the drum center, is agitated by compressed air, and the vacuum used varies from 20 to 23 inches of mercury. The "blow" is used only five minutes per hour.

Oliver filter covers are used with open-weave cloth and brass screens. The life of the filter covers is only 33 days, even with two scrubbings, on account of the lime and calcium sulphate in the water. The filter cake is approximately five-eighths inch thick and contains about 9.8 per cent moisture and 4.4 per cent alumina. A large amount of concentrates is handled per ton of ore concentrated, due to the low concentration ratio, each filter handling about 217 tons of dry concentrates per day. The filter cake drops by gravity directly into railroad cars, and on arrival of the cars at the smelter they are hauled one by one by an electric motor over the arch of a reverberatory furnace, and the content dumped as raw charge into the furnace.

DISPOSAL OF TAILINGS

The tailings are run in a launder 18 inches wide and 9 inches deep with wooden liners to the tailings dam about 1 mile from the concentrator. The launder grade is about five-sixteenth inch per foot. The coarse sand for dam building is separated by spitzkastens along the bottom of the launder, and the sand is delivered by movable iron pipes to the crest of the dam while the finer material is carried by the launder to the far side of the dam and allowed to settle. The launder is carried on trestles which are gradually buried, the bents being made of 2 by 6 inch lumber and spaced 12 feet apart. Material is now being delivered to a new dam, as the old No. 2 dam contains tailings having 0.75 per cent copper, which is reserved for future treatment. This change has resulted in increased cost of tailings disposal.

The clear water from the new dam is pumped to No. 2 dam by a single stage Allis-Chalmers centrifugal pump of 1,200-gallon capacity driven by a 50-hp. motor. The water is elevated 325 feet from No. 2 dam and from the arroyo to the concentrator storage tanks by two 11 by 12 inch Aldrich triplex pumps operated by 125-horsepower motors. One Allis-Chalmers 5 by 4 inch 2-stage centrifugal pump driven at 1,750 revolutions per minute by a 125-hp. motor pumps 1,000 gallons per minute of this water to the concentrator.

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Millions of tons of tailing have been impounded at Cananea during the last 25 years, and not a dam has been lost during all this period. There is no pollution of streams, as only a small amount of alkaline seepage escapes from the dam. Dusting of the tailings has not been noted, as coarse tailings material forms the bank of the dam.

SAMPLING AND ACCOUNTING FOR COPPER

Sampling at Cananea is complicated by the fact that at times ores are received from four or five different mines. These ores must be weighed and sampled separately to credit the various mines for the copper produced.

The ore received from each mine is sampled as a lot each day, and a sample for analysis obtained as shown in Figure 3. The moisture sample, together with the railroad scale weight, gives the dry tons of ore received by the concentrator.

Inventory is taken on the first day of each month, and the total tonnage charged to the concentrator is calculated. This inventory includes estimates of tons of ore in the crude-ore bins and the fine-ore bins and estimates of amount of pulp in the rod mills, ball mills, classifiers, and flotation machines. The tonnage of concentrates on hand in thickeners, filters, clean-up tanks, pounds, and loaded cars is also estimated.

Sampling concentrates

All the concentrates shipped by the concentrator to the smelter each day are called a lot, and the lot is handled as though the concentrator was shipping to a custom smelting plant. Samples of the individual cars of concentrates are obtained at the smelter by the template and auger method and at the concentrator samples of the concentrates are taken on the faces of the filters at intervals of 2 feet. Although smelter and concentrator samples check very closely as a rule the smelter samples are slightly higher in copper than the concentrator samples.

Mill sampling

Samples of rod-mill feed and discharge, classifier sands and overflows, general tailings and concentrates are taken on each shift. Some of the samples are taken for chemical analyses, while others are used in making up monthly composites for screen analyses. The sample of the material discharged from the primary grinding unit is taken to represent the concentrator heads and normally checks closely the monthly average computed from the sampling of the various lots of crude ore. During the rainy season, however, when the wet crude ore sticks in the cars, in the bins, and on the conveyor belts the samples of crude ore tend to give higher copper analyses than the concentrator samples taken as described, thus reducing the concentrator recovery. Samples of general concentrator tailings and concentrates are taken by Galigher automatic samplers. The stream of tailings is cut every 15 minutes by the sampler. From the analyses of samples of heads, concentrates, and tailings, the daily concentrator total and sulphide copper extractions, the iron rejected, and the concentration ratios are calculated for the separate sections and for the entire mill.

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The wet samples are cut to smaller bulk by H. A. Guess sample splitters, filtered in Anaconda pressure filters and dried in a Westinghouse type K electric industrial oven in 12 minutes.

Oxide copper analyses

The method of determining oxide copper consists in treating the sample with 8 per cent sulphuric acid solution at room temperature for 10 minutes with frequent stirring. The quantity of acid solution used is varied according to the amount of pulp taken for the sample. After the 10-minute acid treatment the solution is heated to boiling, filtered, and washed. The copper in the filtrate is determined by standard methods.

WATER SUPPLY

The main water supply for the concentrator is obtained from the Capote mine. Some water is also obtained from the Elisa and Colorads mines, and surplus water from the mine precipitation plant also finds its way into the system. The mine water is very harmful to the flotation process, as it contains various salts, as shown in the analysis of Capote water in Table 14, and also a large amount of organic matter, not shown in the water analysis, derived from the sewage of the mining towns, the drainage of which runs into the concentrator arroyo.

TABLE 14. - Analyses of water from Capote mine before and after lime treatment

Water	Analysis, parts per million							
	Ferrous iron	Ferric iron	Copper	Lime	SO_3	Acid	Alkali P. P.	Alkali M.O
Capote	200	100	92	170	1,170	20		
Treated	Nil	Nil	Nil	1,040	1,440		60	80

Burned lime is added to the arroyo water as it flows by the concentrator, precipitating the gelatinous material and eliminating the deleterious elements as much as possible. Tailings are also added to settle and bury the precipitate. The lime is added at the rate of 8.8 pounds per ton of ore, of which 5.0 pounds per ton is used to give the desired alkalinity for flotation and 3.8 pounds per ton for water treatment. The lime is burned locally by the company at a cost of $5.50 per ton and has the following analysis:

CaO	Insoluble	Iron	Available CaO
84.5	3.6	0.8	84.4

The analysis of the water after lime treatment is shown in Table 14. Although the iron salts are removed completely by the lime treatment, the water is saturated with calcium sulphate, which causes trouble and expense in the flotation department. The treated water is pumped from the tailings dam to the concentrator water-storage tanks, as described under "Disposal of tailings."

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CONCENTRATOR RECOVERY

Table 15 gives the tonnages and analyses of the ores treated by the concentrator during the first half of the year 1929. Table 16 gives the weights of copper, silver, and gold received by the concentrator during the same period.

TABLE 15. - Weights and analyses of ores treated

Ore	Weight, dry tons	Per cent of total ore	Analyses: Copper, per cent	Silver, ounces per ton	Gold ounce per ton	Iron, per cent	Sulphur, per cent	Lime, per cent	Alumina, per cent	Silica, per cent
Colorada	233 494	57 51	4.33	0.49	0.019	7.5	6.7	0.2	11.6	58.4
Capote	136,995	33.74	1.78	.40	.004	15.9	14.6	.4	10.7	44.9
Veta	35,094	8.64	2.22	.19	.003	7.0	6.4	.2	7.6	64.6
Elisa	426	.11	2.82	1.86	.016	21.0	9.5	9.6	7.4	26.4
Total and average	406,010	100.00	3.29	.43	.013	--	--	--	--	--

TABLE 16. - Weights of copper, silver, and gold received by the concentrator

	Weight of copper, pounds	Weight of silver, ounces	Weight of gold, ounces
Colorada	20,216,908	113,367	4,529
Capote	4,888,352	54,170	496
Veta	1,560,948	6,974	92
Elisa	24.036	793	7
Total	26,690,244	175,126	5,124

Table 17 gives the weights and analyses of concentrates produced from the various ores during the first half of the year 1929, and Table 18 gives the weight and analysis of the tailings produced during the same period. Table 19 shows the plant recovery of copper, silver, and gold.

TABLE 17. - Weights and analyses of concentrates

Ore	Weight, dry tons	Copper: Per cent	Copper: Pounds	Silver: Ounces per ton	Silver: Ounces	Gold: Ounces per ton	Gold: Ounces
Colorada	41,232	23.76	19,593,398	2.64	108,913	0.097	2,995
Capote	14,564	15.43	4,495,716	3.59	52,235	.030	441
Veta	3,744	19.25	1,441,694	1.75	6,557	.022	80
Elisa	47	24.34	23.228	15.69	748	.122	6
Total	59,589	21.44	25,554,036	2.83	168,455	.076	4,524

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TABLE 18. - Weight and analysis of tailings

	Weight, dry tons	Copper		Silver		Gold	
		Per cent	Pounds	Ounce per ton	Ounces	Ounce per ton	Ounces
Tailing	346,421	0.19	1,289,210	0.03	10,923	0.002	600
Unaccounted for			-153,002		-4,251		
Corrected tailings	346,421	.16	1,136,208	.02	6,671	.002	600

TABLE 19. - Copper, silver, and gold recoveries, per cent

	Copper	Silver	Gold
Recovery	95.74	96.19	88.28

Tables 20 and 21 give screen analyses of typical concentrates and tailings, respectively, with chemical analyses of the various screen sizes. Table 22 gives a microscopic analysis of the copper losses in the tailings.

TABLE 20. - Screen analysis of concentrates

Mesh	Weight, per cent	Analysis, per cent				
		Total copper	Sulphide copper	Oxide copper	Iron	Sulphur
On 48	1.30	29.14	29.09	0.05	26.6	31.1
On 65	4.74	26.62	26.58	.04	24.8	30.9
On 100	11.52	24.78	24.73	.05	24.4	31.3
On 150	12.00	24.14	24.09	.05	26.4	34.1
On 200	9.00	25.10	25.05	.05	27.2	35.3
On 250	14.30	26.10	25.94	.16	28.4	36.4
On 325	8.58	27.86	27.62	.24	27.2	35.5
Through 325	38.56	24.58	23.63	.95	19.4	25.8
Total and average	100.00	24.77	24.38	.39	23.6	29.6

TABLE 21. - Screen analysis of tailings

Mesh	Weight, per cent	Analysis, per cent				
		Total copper	Sulphide copper	Oxide copper	Iron	Sulphur
On 48	1.64	0.72	0.69	0.03	2.4	1.5
On 65	7.40	.41	.38	.03	2.2	1.5
On 100	13.42	.27	.24	.03	4.0	3.7
On 150	10.60	.15	.13	.02	8.6	9.7
On 200	6.72	.14	.11	.03	11.4	11.7
Through 200	60 22	.10	.08	.07	6.2	5.8
Total and average	100 00	.16	.11	.05	6.2	5.9

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TABLE 22. - Microscopic analysis of copper losses in tailings

	Analysis, per cent copper	Copper in tailings, per cent	
		Sulphide copper	Total copper
Free copper minerals			
Chalcopyrite	0.02475	17.24	13.26
Bornite	.00104	.73	.56
Chalcocite	.10594	73.81	56.74
Covellite	.00451	3.14	2.42
Total copper as free mineral	.13624	94.92	72.98
Copper minerals attached to quartz			
Chalcopyrite	.00279	1.95	1.88
Bornite	.00068	.48	1.49
Chalcocite	.00351	2.44	.16
Covellite	.00030	.21	.36
Total copper as included, grains	00728	5.08	3.89
Total sulphide copper	.1435	100.00	76.87
Oxide copper	.0432		23.13
Total copper	.1867		100.00

Table 23 gives a summary of concentrator costs for the first half of the year 1929.

TABLE 23. - Summary of costs, in cents per ton of ore concentrated

	Operating				Repairs		Miscellaneous	Total
	Labor	Power[1]	Supplies	Reagents	Labor	Supplies		
Coarse breaking to 3-1/2 inch size								
Intermediate crushing to 1-1/2 inch size	1.87	1.40				0.46		3.73
Primary grinding to 10-mesh size	.73	3.00	2.40			.12		6.25
Secondary grinding to 48-mesh size	.29	6.21	4.57			.52		11.59
Classifying, screening, and conveying	.25	.99				.04		1.28
Flotation	1.88	4.96		5.54		.72		13.10
Dewatering and handling concentrates	.71	.51				.32		1.54
Assaying and sampling	.69							.69
Tailings disposal	.76							.76
Water supply	.79	2.65				.48		3.92
Miscellaneous	75	.62	4 06		3 22	6 35	3.87	18 87
Total	8 72	20 34	11 03	5.54	3 22	9 01	3 87	61 73

1 - Power costs, 1.3 cents per kilowatt hour.

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Table 24 gives a summary of labor, and Table 25 gives the distribution of power for the first half of the year 1929.

TABLE 24. - Labor summary

	Operating	Repairs	Total
Ore treated per 8-hour shift, tons	22 54	76 73	15.90

TABLE 25. - Power distribution

	Kw.h. per ton	Per cent of total power
Breaking and crushing	1.051	6.9
Primary grinding	2.685	17.6
Secondary grinding	4.666	30.6
Classifying, screening, and conveying	.307	2.1
Flotation	3.719	24.3
Dewatering and handling concentrates	.386	2.5
Disposal of tailings	-	-
Water supply	1.992	13.0
Miscellaneous	.464	3 0
Total	15.270	100 0

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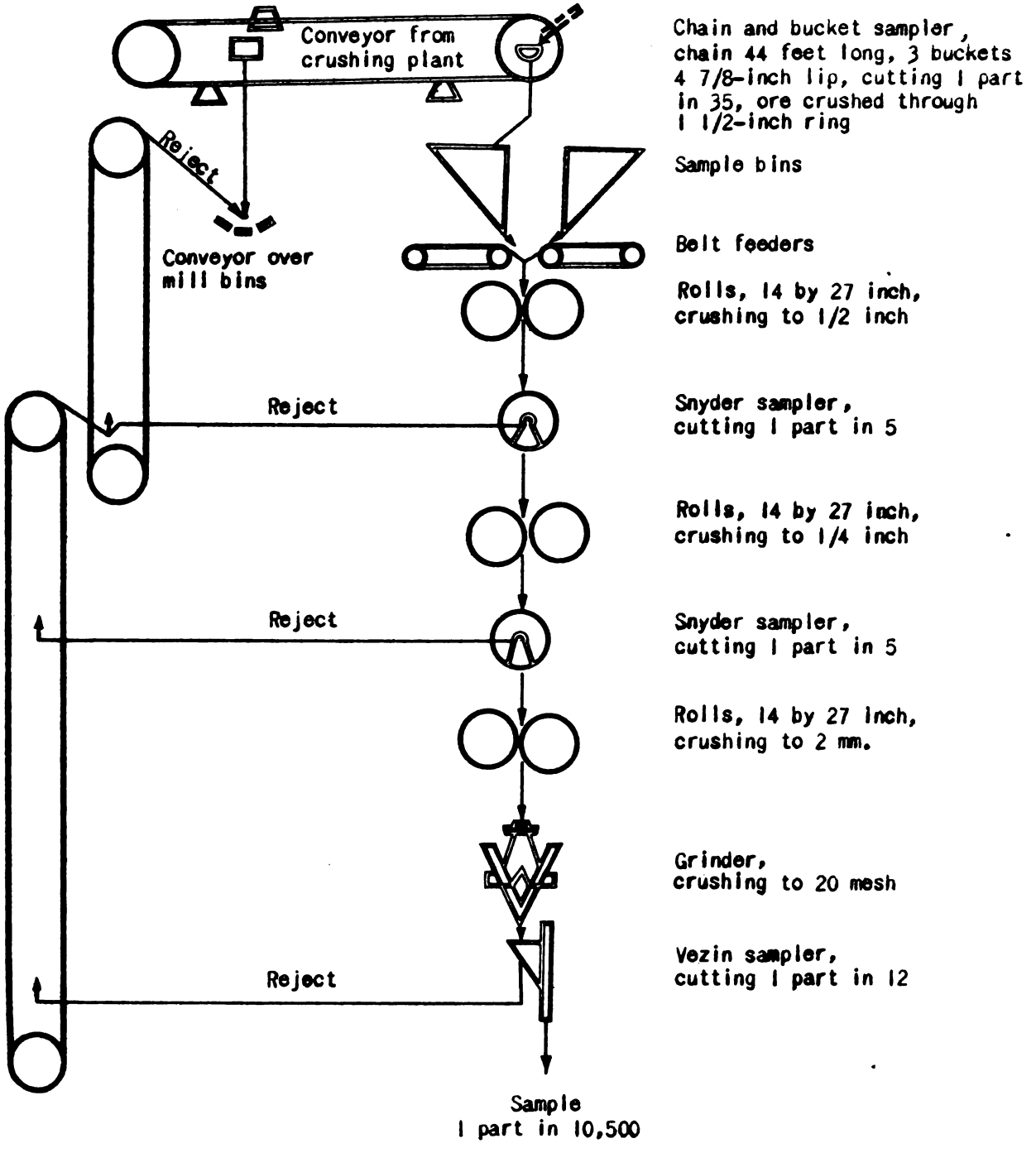

Figure 3. - Flow sheet of sampling plant

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